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November 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

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MINING AND MILLING METHODS AND COSTS
HOG MOUNTAIN GOLD MINING & MILLING CO., ALEXANDER CITY, ALA.

BY

N. O. JOHNSON

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MINING AND MILLING METHODS AND COSTS, HOG MOUNTAIN GOLD MINING & MILLING CO., ALEXANDER CITY, ALA.[1]

By N.O. Johnson[2]

CONTENTS

	Page
Introduction	2
Acknowledgments	2
Mining	2
History	2
Topography and climate	3
Geology	4
Physical characteristics of ore and wall rocks	5
Prospecting and exploration	6
Sampling and estimation of tonnages and values	6
Development	7
Stoping	7
Tramming and hoisting	8
Ventilation and drainage	8
Air compression	9
Steel sharpening	9
Miscellaneous	9
Costs	10
Milling	11
General description	11
Ore treated	12
Present method of concentration	12
Coarse crushing	12
Picking	12
Fine crushing	13
Grinding and classifying	14
Flotation	15
Thickening, filtering, and drying	15
Mill control	16
Costs	18
Tables	20

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6914."

[2] Mining Engineer, Hog Mountain Gold Mining & Milling Co. and one of the consulting engineers, U.S. Bureau of Mines.

INTRODUCTION

This is one of a series of Information Circulars dealing with mining and milling methods and costs at mines in various districts in the United States

The Hog Mountain gold-mining district is at the northern edge of Tallapoosa County, in the east central part of the State of Alabama. The district is equally close to Goodwater and to Alexander City, both stations on the main line of the Central of Georgia Railroad.

From Alexander City the mine is reached by 13 miles of graded road, traveling in a northeast direction. In dry seasons the road is good; in wet weather it is bad but passable. Improvements are being made constantly on the road, so that in the near future the gravel road should be good under all weathered conditions.

Hog Mountain lies in the mineralized zone that extends northeasterly through Georgia into the Carolinas and Virginia. The sulphide ores in this district are in the very early stage of development, hence present methods may be changed considerably in the future, but enough work has been done at the property of the Hog Mountain Gold Mining & Milling Co. so that its methods should be of general interest.

ACKNOWLEDGMENTS

To the directors and George M. Brown, manager, of the Hog Mountain Gold Mining & Milling Co., thanks are given for permission to present this paper for publication.

Acknowledgment also is made to C.F. Park, Jr., of the United States Geological Survey, for information used in the section on the geology of the mine and the district, and to E.J. Alderfer, Jr., and R.M. Fuller, both members of the staff, for their able assistance in furnishing constructive criticism and compiling cost data.

MINING METHODS

History

Gold was first discovered at Hog Mountain in 1839 by a prospector named Johnston. First operations were carried on in 1844 and 1845, during which time the ore was mined and milled with the crudest appliances, and the ore was hauled by oxen. During these years a 10-stamp mill and amalgamation plates were used to recover the gold; but production was exceedingly small, and it is estimated that less than 1,000 tons were mined and milled at that time.

After this first brief operation the property remained idle until 1896, when it was purchased by T.H. Aldrich, who started mining the oxidized surface ore along the numerous veins. A cyanide leaching plant to treat 75 tons per 24-hour day was constructed, and between 1896 and 1916 approximately 100,000 tons were treated. During this period most of the veins on the northern half of the mountain were developed. The veins on the southern end of the granite plug were too distant from the mill at that time to warrant much development work.

Mining was carried on to a depth corresponding to the line of demarcation between the oxidized and sulphide ores. When this depth was reached (20 to 60 feet), the individual workings were abandoned and another vein was opened up. A shallow crosscut tunnel approximately 600 feet long was driven to a point approximately 1,000 feet south of the old mill. This tunnel was instrumental in exposing the numerous parallel veins that exist in this area. Due to higher labor and material costs during the war and to the depletion of the more amenable oxidized ores the mine was abandoned in 1916.

After this second period of operation, the property remained idle until 1931, when E.H. Emerson made a complete report on the property. Shortly afterward, serious exploration began under the direction of P.S. Gardner and George M. Brown. From then until the present, work has consisted of drifting and stoping along the Blue and Barren veins and a very small amount of development work on other veins.

The surface construction work included the erection of a 100-ton flotation mill and the building of an assay office and blacksmith, machine, and carpenter shops, together with the necessary living quarters. Water for all purposes is pumped directly from a small clear stream 1 mile from the camp through a 4-inch cast-iron pipe to a 16,000-gallon wood tank on the mountain side 120 feet above the elevation of the ball-mill floor. Necessary power is obtained by a 12.6-mile, 44,000-volt transmission line connected to the Alabama Power Co. main line at Alexander City.

The mill was started in February 1934, and, although production has been relatively small for gold mines working in this grade of ore, as compared to most other gold properties in southeastern United States, production has been large.

Topography and Climate

Hog Mountain lies within the Ashland Plateau. It forms a conspicuous ridge rising some 300 or 400 feet above the surrounding country and reaching a height of more than 1,000 feet above sea level. The topography is typical of the gently rolling, deeply eroded Southern Appalachian region. Hillabee Creek, a branch of the Tallapoosa River, constitutes the largest part of a good drainage system.

The climate is mild and humid. The mean rainfall is approximately 55 inches, being well distributed throughout the year. Vegetation typical of the region includes hickory, oak, and long- and short-leaf pine. Most of the trees are second growth, but a few areas of uncut timber remain. A thick undergrowth covers most of the country.

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Geology

The Wedowee formation and the intrusive quartz-diorite, considered to be an offshoot of the much larger areas of Pickneyville granite to the west, have been distinguished at Hog Mountain.

The Wedowee formation consists of fine-grained graphite schists. The schists are of sedimentary origin in part at least, and in places are partly recrystallized, containing mostly quartz, amorphous graphite, and fine-grained sericite. Bedding planes are almost obliterated, but where present are cut by the schistosity at an angle. The schist planes are intricately folded and faulted but generally strike north 10 to 20 degrees east and dip steeply eastward. The age of the formation has not been definitely established.

The intrusive quartz-diorite is light gray, with an even granitoid texture. It is in this formation that the auriferous veins occur. It covers an area more than 4,500 feet long and about 1,000 feet wide. (Fig. 1.) The strike of its long axis is roughly parallel to the schistosity of the surrounding schists. The majority of the veins strike between N. 60 E. and due east-west, with dips of 45 to 85 degrees to the north. In a few places where followed toward the schist and quartz-diorite contact the veins widen, forming pipe-shaped ore bodies, but commonly they split into many stringers and pinch out at the contact. Shear zones, conspicuously developed in the quartz-diorite, are less noticeable in the schists. They strike almost due east and dip northward. Many shear planes cross the veins, though some bend and merge with them. Certain shear zones, while carrying less value than the vein material proper, are still high enough in grade to be economically mined and milled.

Erosion in the neighborhood of Hog Mountain has been considerable, and a great deal of ore must have been lost as the deposits, which are of the deep-vein type, must have extended up into the eroded portion. Had the gold been coarse, this erosion would have resulted in placers of real importance.

The former operators report that the ore was slightly richer at the gradation between the oxide and sulphide zones. Some concentration of value has been found near open-cuts by the present operators. As there is small difference between the assays from the present workings in sulphide ore and the reported assays from the old workings in the oxidized ore, it is evident that supergene enrichment has been unimportant.

Mineralogy. - Quartz is the most abundant vein mineral. Below the water level it is generally massive, vitreous, dark or bluish-gray where associated with gold, and usually light gray or white where barren. The gold values are associated with the sulphides. Pyrrhotite is the most abundant sulphide, while chalcopyrite, arsenopyrite, sphalerite, and pyrite occur, but usually in small amounts. All the gold is finely divided and is intimately mixed with or enclosed in the sulphides. Free gold can easily be distinguished in the flotation concentrates under the microscope, and it is possible but extremely difficult to obtain very fine colors by panning. Silver is present but to the extent of hardly more than a trace. Flotation concentrates carry about 0.4 ounce of silver to the ton.

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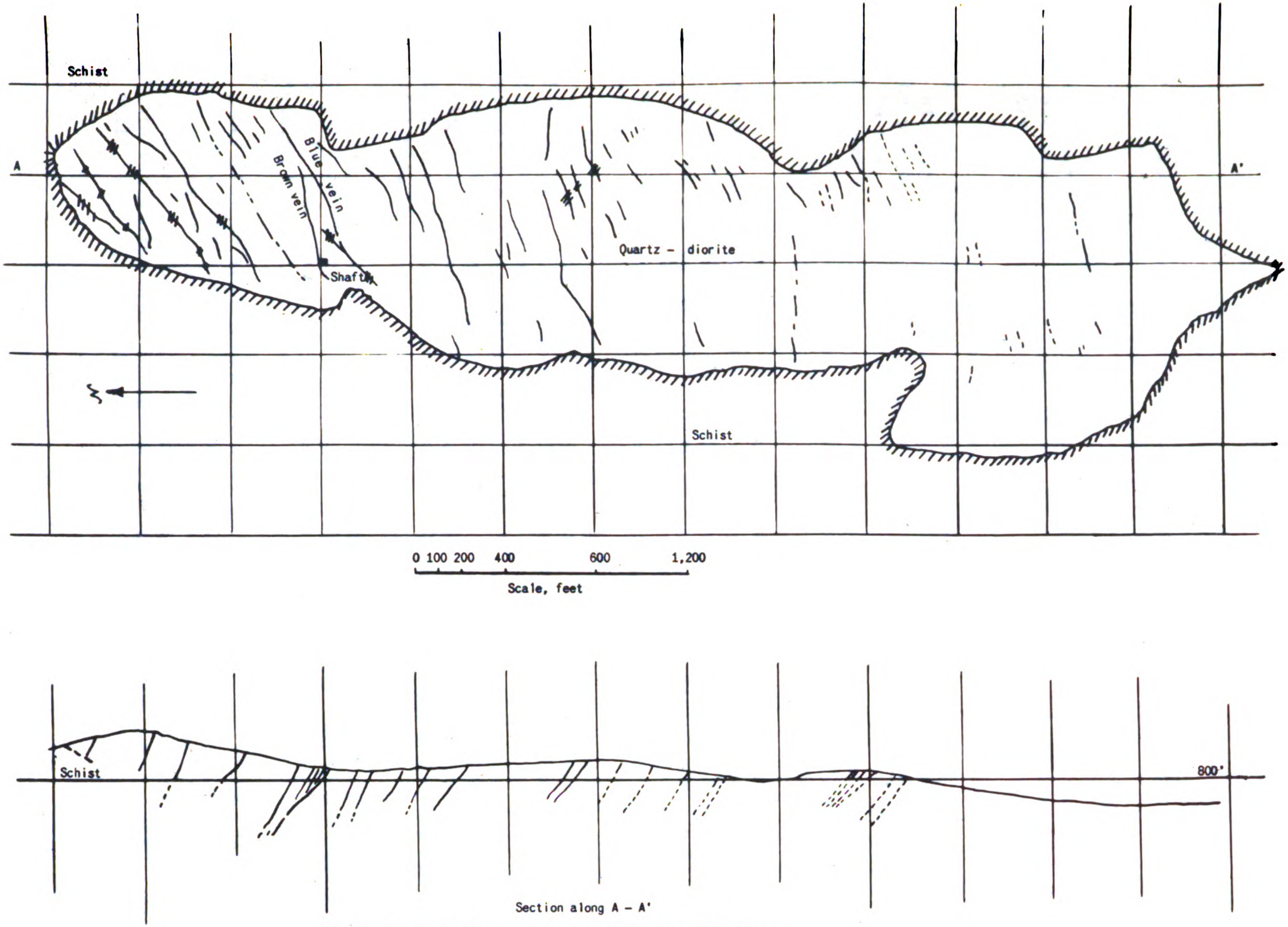


Section along A - A'

Figure 1.- Geologic map of Hog Mountain intrusive mass.

Physical Characteristics of Ore and Wall Rock

The ore has been mined by shrinkage stoping. The important factors that govern the choice of the mining method are as follows:

1. The walls are normally strong and well-defined, but in zones of intense shearing the country rock is highly silicified and mineralized for 1 to 4 feet beyond the boundaries of the true quartz vein. In these sheared areas the walls are irregular.

2. In areas where the shearing is either parallel or at an angle to the veins slabbing off of the hanging wall does not cause any difficulty, while stoping for the broken ore offers the needed support. However, in drawing the ore from the chutes large slabs of wall rock are drawn into the chutes with the ore, where they cause trouble in drawing and dilute the ore. This trouble from shearing in the adjacent walls has been experienced in only one of the four completed stopes. Future stoping of such areas will either be by timbered stopes or else ore pillars will be left in where most needed.

3. The length of the stopes in which the shrinkage system is used is limited because of the large tie-up of ore from the time the stope is started until it is completed. The approximate stoping length is now 100 feet, which under good conditions can be finished in 60 days. The reason for limiting the length of stopes is that delayed pulling causes serious oxidation of the broken ore. Under the present metallurgical practice, this is detrimental to a good mill recovery.

4. The veins dip 45 to 85 degrees. Forty-five degrees is not steep enough to allow a uniform drawing of the ore, particularly if the footwall is irregular. Under these conditions at least a third of the ore lies on the footwall of the stopes between the chutes (20 to 25 feet apart). This ore is extracted either by shoveling in the stopes or, preferably, by using an air hoist with a dragline attachment. Any slabs that are attached from the hanging wall fall into the stope, which makes shoveling dangerous and drawing of the ore difficult.

5. Because of the irregular tenor of the ore it is difficult to supply ore of uniform grade to the mill; shrinkage stoping requires a close check on the daily ore broken, for the ore may be broken several weeks in advance of its reaching the mill.

6. Since the veins are mined only in areas where the widths are greater than 3.5 feet, there is little chance for dilution from the wall rock, unless either shearing or a roll occurs in the stope. In these cases the wall rock is slightly weaker, and pieces of the surrounding rock are drawn into the cars along with the broken ore. Shrinkage stoping prevents any underground sorting, which would be an advantage in such cases.

Prospecting and Exploration

Prospecting and exploration are done by drifting and raising; prospect headings frequently are run from the stopes. As the ore cannot be projected too far ahead of actual exposures, exploration workings follow the ore as closely as possible.

Diamond drilling from an underground station has been tried recently, and although a good core of the structure drilled can be obtained, the extreme hardness of the rock wears the bits excessively. Bortz bits, the type first used, became polished after as little as 17 feet of drilling. Black-carbon bits will be used for any future drilling that may be done, and although no tests have been made it is expected that results will be better.

Frequently holes are drilled into either the footwall or the hanging wall of the vein with long pieces of steel from a drifting set-up position and help to locate the bodies of quartz in parallel vein structures.

Crosscutting the whole vein structure is the only adequate method of exploring any section of the vein, for, from the amount of work accomplished in the past the vein structure apparently consists of a great number of parallel veins and stringers.

It is planned to explore the veins north of the present workings from the shaft to determine the position of the east and north contact of the quartz-diorite plug and to explore the veins below the present workings.

Sampling and Estimation of Tonnages and Values

Considerable sampling is done, both while exploring and while stoping, to control the grade of the material delivered to the ore bin. All muck piles in the face are sampled after each round. The assays of these samples determine whether the muck is sent to the ore bin or to the waste dump.

Assaying is done by the mine assayer and one assistant, who run about 50 samples daily, of which most are mill samples.

Longitudinal maps of all stopes are made, from which the broken ore reserves are estimated. The grade of these reserves is estimated from the grab samples taken in the course of stoping and is checked by the grab sampling of all the cars drawn from each stope. Of the five major stopes completed, the average width is 6 feet and the length 140 feet.

At the end of each month the stopes and the development workings are measured and plotted on the mine maps. The volume of the broken ore is estimated by multiplying the area of the ground stoped along the plane of the vein by the average of the width of the vein measured at various intervals. The volume of broken ore is divided by 20 to obtain the tonnage, while a factor of 12 cubic feet per ton is used for rock in place.

Using the workings in the shaft as a basis for calculations, 2,900 feet of drifts have been run along the veins, from which approximately 70,000 tons of ore have been mined or developed, including underground stoping done on these veins in former days (not including surface oxidized ore).

Development

The mine is worked through a two-compartment shaft 215 feet deep, 200 feet east of the west contact of the quartz-diorite plug and 1,200 feet from the north end. At 160 feet below the collar of the shaft the west contact is encountered, which indicates that the formation dips less than 45 degrees at this particular point.

Each compartment of the shaft is 4-1/2 by 6 feet in the clear, and is timbered with 8- by 8-inch sets at 5-foot centers. One compartment is used for the cage while the other is employed for the air and water lines, electric cables for lighting the stations and signals, and a ladderway.

Stations are cut 10 feet high by 15 feet wide at the shaft and taper to drift size in 20 to 30 feet. Stations are cut on both sides of the shaft.

All drifts and crosscuts are 5 by 7 feet in size, and nearly all are untimbered. Drilling is done with 3-1/2 inch drifting machines mounted on vertical columns, using 1-1/4 inch hollow round steel. Standard crossbits with double taper, gage changes of 1/8 inch, and length changes of 12 inches are used. A round consists of 13 to 16 holes, depending on the nature of the ground. Most economical results to date have been obtained with Hercomite blasting dynamite, and an average advance of 4 feet per shift can be maintained in each heading.

All mucking is done by hand into 16- and 20-cubic foot, box-type, end-dump cars.

Raises are driven as needed for the shrinkage-stope draw points and for ventilation. A considerable amount of prospect work also is done by raising. A round in the raises is drilled by one miner and a helper using a self-rotating wet stoper and 1-inch hollow hexagon steel.

Stoping

Distribution of the gold within the ore body is erratic; small and large horses of granite are encountered, and the trend of the ore is so irregular that no stope layout can be designed ahead of the stoping. Stoping limits are determined by the width and the assay.

It is impossible to estimate accurately the grade of the ore by eye; therefore, sampling is relied upon to guide the stoping. Experienced men, however, can distinguish barren or low-grade vein material from that which contains workable values.

The shrinkage method of stoping is used because of the excellent walls and the fair widths of stopes.

Drilling in the stopes is done with the same machines and steel that are employed in the raises. Holes are drilled 4 to 8 feet deep and 2 to 4 feet apart. However, where the values go into the wall of the stopes, as is true in zones of shearing, slabbing rounds using a drifter are occasionally necessary. Blasting is done with 60-percent dynamite and electric delay caps when a large number of holes are to be fired. Usually the whole back is drilled and then holes fired simultaneously. This practice minimizes fumes in the stope. Wherever possible the work is done under contract. In drifting the men are paid per foot of advance while in the stopes per foot of hole drilled.

In places where the vein dips less than 45 degrees a compressed-air hoist has been employed to advantage to operate a scraper for cleaning the footwall of the stopes.

Manways into the stopes have a single compartment, 3-inch cribbing being used along stulls set about 5 feet apart. Chutes are made of timber and are spaced 15 to 20 feet apart. Usually rock pillars are left over the drifts between chutes, but in some instances the broken ore is carried on tunnel sets over the drift. An effort is made to reduce all the boulders in the stopes to 12 inches or smaller before being drawn into the muck. In spite of this precaution, there are some that must be blasted in the chutes.

Tramming and Hoisting

All tramming from the stopes is done by hand. The average distance from the stopes to the stations is about 350 feet. Switches are placed at each station, which reduces time loss when the cage returns with an empty car. The loaded cars are hoisted from the 100- and 200-foot levels to the surface, where an empty car waits on a switch to replace the loaded car taken off. The total elapsed time from the loading of the cage to its return to the 200-foot level with an empty car is 2-1/2 minutes. The surface trammers attend to the changing of the cars on the cage and tram the loaded cars to the bin at the head of the jaw crusher, which is 75 feet from the shaft.

A single-drum steam hoist, which has been converted to electric drive, is powered by a 50-horsepower, 220-volt induction motor through double reduction gears.

Ventilation and Drainage

The workings in and near the shaft are ventilated naturally; this has been made possible by connecting two stopes with the surface. However, in dead-end workings the ventilation is poor necessitating running a motor-driven fan on the surface. The air is delivered through 12-, 10-, and 8-inch galvanized pipes.

The workings are virtually dry. A sump 15 feet deep acts as a reservoi for all the shaft drainage. It is pumped out twice a day by a centrifugal pump, direct connected to a 10-horsepower, 3,400-r.p.m. motor. The pump handles 150 gallons per minute and empties the sump in approximately 2 hours.

Air Compression

Compressed air is supplied by two compressors. One is a duplex, two-stage type, 17 by 10 by 12 inches, and is driven by a 14-inch endless belt from a 150-horsepower, 2,200-volt, induction motor; the other is an angle-compound, 17 by 9-3/4 by 12 inches, and is driven by a 14-inch endless belt from a 150-horsepower, 2,200-volt, induction motor. The combined capacity of the two machines is approximately 1,500 cubic feet of air per minute.

Steel Sharpening

The blacksmith shop has one Ingersoll-Rand type no. 50 and one Denver rock drill Model no. 8 sharpener. Other equipment is an oil furnace, a compressed-air emery wheel, and a forge. Two shifts handle a maximum of 900 pieces of steel daily in addition to making up new steel and doing other necessary blacksmith work.

Miscellaneous

A complete machine and carpenter shop aids substantially in construction and repair work.

Electric power is delivered to the mine substation, where six 75- kv.-a, 44,000-2,200 volt transformers are located. Large motors operate on 2,200 volts, while the remainder of the power is taken at 220 and 110 volts from three additional 75-kv-a, 2,300-200/100-volt transformers. The total connected horsepower (name-plate rating) on the property is 877-1/2, with kilowatt demand of 359.2. The power consumption is between 150,000 and 200,000 kw.-h monthly.

Water for all purposes is delivered to a 16,000-gallon tank near the mill. A centrifugal pump, operating at 3,500 r.p.m. and driven by a 40-horsepower motor, pumps the water through a 5,300-foot 4-inch cast-iron pipe against a 300-foot vertical head.

The assay office has all the equipment necessary for analytical and experimental work. Assays from underground workings are plotted on the various maps - one longitudinal view for each vein and a plan view of each level.

Costs

TAPLE 1. - Summary of costs

Hog Mountain Gold Mine

7-month period
(October 1935 - April 1936)

Ore hoisted and milled during period - 14,453 tons.
Mining method - shrinkage stoping.

	Mining Costs per Ton of Ore				
	Labor	Power	Explosives	Materials	Total
Breaking ore:					
Miners	$0.348	-	-	-	$0.348
Explosives	-	-	$0.239	-	.239
Machine drilling	.064	$0.225	-	$0.033	.322
Cost of breaking ore	.412	.225	.239	.033	.909
Tramming and mucking	.189	-	-	.047	.236
Timbering	.091	-	-	.036	.127
Hoisting and dumping	.050	.064	-	.013	.127
Assaying and sampling	.019	.010	-	.020	.049
Miscellaneous expense	.078	.038	-	.040	.156
Total	.839	.337	.239	.189	1.604

Note. - The total cost of breaking ore is derived from 18,093 tons broken in the shrinkage stopes during this period, and the total cost in the table is that of the stoped ore delivered at the mill bin.

Development costs have been proportionately high due to need for increasing the number of stope faces available to meet demands for increased production. A recent study of all the development to date indicates the normal development cost will be 50 to 75 cents per ton of ore milled. On a footage basis, drifts and crosscuts average $8.55 per foot and raises $11.45 per foot, including all items of cost covered in Table 1 for stope ore.

TABLE 2. - Summary of costs in units of labor, power, and supplies

Hog Mountain Gold Mine
7-month period
(October 1935 - April 1936)

Ore hoisted and milled during period - 14,453 tons.
Mining method - shrinkage stoping.

	Mining (stoping)
Labor (man-hours per ton of ore hoisted):	
Breaking ore (drilling and blasting)	1.58
Tramming and mucking	1.00
Timbering	.18
Hoisting and dumping	.22
Assaying and sampling	.06
Other expenses	.31
Total labor underground	3.35
Average tons per man-shift underground	2.68
Labor, percent of total cost	50.2
Power and supplies:	
Explosives (Hercomite), pounds per ton	1.5
Timber, board feet per ton	1.2
Total power, kilowatt-hours per ton	34.0
Compressing	26.0
Hoisting	4.7
Pumping and ventilation	3.3

Note. - Under timber, all tree timber cut on the property and used for stulls, etc., is not measured and therefore is not included in the board-feet-per-ton figure. The labor used to obtain these trees is charged in the mine-timbering labor account.

MILLING

General Description

Milling operations were started at the Hog Mountain property in 1893, when a cyanide leaching plant was treating coarsely ground (minus 1/4-inch mesh) oxidized gold ores. After approximately 100,000 tons of this ore had been milled there remained a very small ore reserve economically subject to treatment. Experiments were then started on the less-limited supply of sulphide ores. However, with war prices and scarcity of labor prevailing at that time, together with the unfavorable experimental results obtained from the sulphides, the mine and mill were closed in 1916. The property remained idle until 1933, when extensive sampling and metallurgical tests were made on the sulphide ores. Some of these tests were conducted locally and others by the American Cyanamid Co. of New York, and after a complete study of the ore to be handled it was finally decided to use straight flotation with possible cyaniding of the concentrates after the tonnage of the mill had been increased sufficiently.

Ore Treated

The Hog Mountain ore consists mainly of quartz containing pyrrhotite and pyrite as the principal metallic minerals with small amounts of chalcopyrite. The quartz-diorite wall rock in zones of shearing has small stringers of these same minerals and such places are taken as ore. The gold is associated with these sulphides, the average gold content of 46,541 tons milled in the last 26 months being 0.184 ounce per ton.

Present Method of Concentration

A flow sheet showing the general arrangement of the machines is presented in figure 2.

A general summary of method of treatment follows:

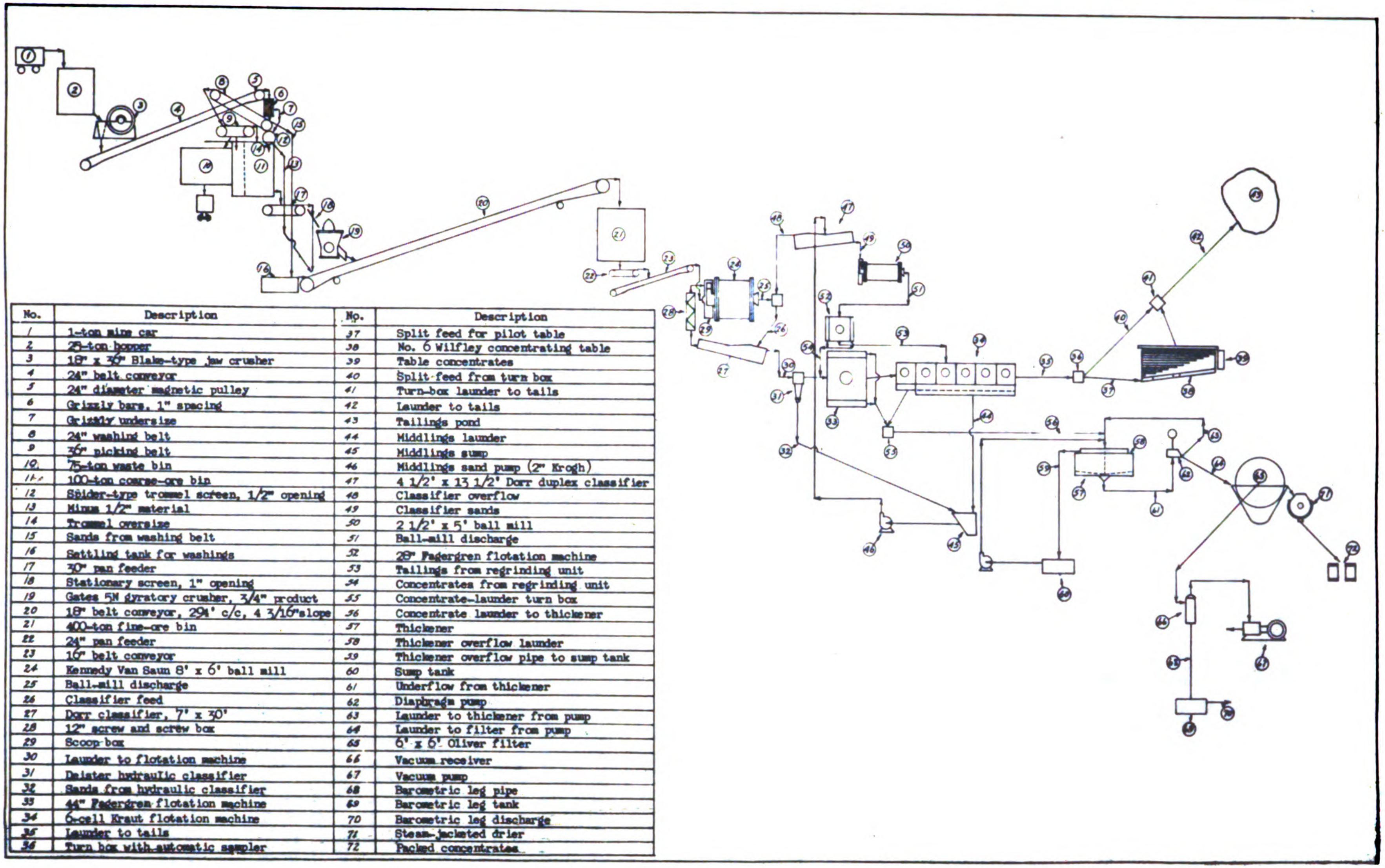
The ore, coming directly from the mine cars, is crushed to 4 inches in an 18- by 36-inch jaw crusher. After passing over a picking belt, the ore is crushed in a 5N Gates gyratory crusher, then delivered to an 8- by 6-foot ball mill operated in closed circuit with a Dorr classifier. The classifier overflow is floated, giving a 3-ounce concentrate, while the middling product is returned to the mill for regrinding. The concentrates are thickened, filtered, and dried and then packed in drums for shipping. The tailings, after passing over a pilot table, are disposed of as waste.

Coarse Crushing

Mine-run ore is delivered to a 25-ton hopper 75 feet west of the present 200-foot shaft. Double tracks lead from the shaft to this hopper, thus reducing time lost while dumping. The ore from this hopper gravitates into an 18- by 36-inch Blake-type jaw crusher. A gate in the chute is operated by the crusherman and prevents any choking of the crusher or overloading of the picking belt. The crusher is driven by a 75-horsepower, 2,200-volt, slip-ring motor through a 12-inch belt drive at 230 r.p.m. The discharge opening is set at 4 inches and, with a 3/4-inch stroke, has a capacity of approximately 50 tons per hour.

Picking

From the jaw crusher the ore drops onto a 24-inch belt conveyor inclined 21 degrees from the horizontal and which is 78 feet long center to center of pulleys. The belt is driven from the main-line shaft in the picking building at a speed of 106 feet per minute. The ore, after passing over a magnetic pulley, drops onto a grizzly 3 feet wide by 8 feet long, inclined at 40 degrees, and having the bars spaced to 1-inch openings. After the fine material has been screened out the oversize drops onto a washing belt equipped with high-pressure water sprays placed directly over the belt. This belt is 24 inches wide and 21 feet long center to center of pulleys, and moves 43 feet per minute at a 22-degree incline from the horizontal. The wash water carrying the fine sands is collected at the tail pulley of this same belt and transferred to a settling tank at the discharge of the fine crusher. Intermittently the settled sands are shoveled on to the main conveyor belt to the mill bin. After being washed the coarse material drops onto a sorting



No.	Description	No.	Description
1	1-ton mine car	37	Split feed for pilot table
2	25-ton hopper	38	No. 6 Wilfley concentrating table
3	18" x 36" Blake-type jaw crusher	39	Table concentrates
4	24" belt conveyor	40	Split feed from turn box
5	24" diameter magnetic pulley	41	Turn-box launder to tails
6	Grizzly bars, 1" spacing	42	Launder to tails
7	Grizzly undersize	43	Tailings pond
8	24" washing belt	44	Middlings launder
9	36" picking belt	45	Middlings sump
10	75-ton waste bin	46	Middlings sand pump (2" Krogh)
11	100-ton coarse-ore bin	47	4 1/2' x 13 1/2' Dorr duplex classifier
12	Spider-type trommel screen, 1/2" opening	48	Classifier overflow
13	Minus 1/2" material	49	Classifier sands
14	Trommel oversize	50	2 1/2' x 5' ball mill
15	Sands from washing belt	51	Ball-mill discharge
16	Settling tank for washings	52	28" Fagergren flotation machine
17	30" pan feeder	53	Tailings from regrinding unit
18	Stationary screen, 1" opening	54	Concentrates from regrinding unit
19	Gates 5N gyratory crusher, 3/4" product	55	Concentrate-launder turn box
20	18" belt conveyor, 294' c/c, 4 3/16"slope	56	Concentrate launder to thickener
21	400-ton fine-ore bin	57	Thickener
22	24" pan feeder	58	Thickener overflow launder
23	16" belt conveyor	59	Thickener overflow pipe to sump tank
24	Kennedy Van Saun 8' x 6' ball mill	60	Sump tank
25	Ball-mill discharge	61	Underflow from thickener
26	Classifier feed	62	Diaphragm pump
27	Dorr classifier, 7' x 30'	63	Launder to thickener from pump
28	12" screw and screw box	64	Launder to filter from pump
29	Scoop box	65	6' x 6' Oliver filter
30	Launder to flotation machine	66	Vacuum receiver
31	Deister hydraulic classifier	67	Vacuum pump
32	Sands from hydraulic classifier	68	Barometric leg pipe
33	44" Fagergren flotation machine	69	Barometric leg tank
34	6-cell Kraut flotation machine	70	Barometric leg discharge
35	Launder to tails	71	Steam-jacketed drier
36	Turn box with automatic sampler	72	Packed concentrates

Figure 2.– Flow sheet of flotation mill, Hog Mountain Gold Mining & Milling Co.

belt 36 inches wide and moving 27 feet per minute. At this speed the three pickers stationed along the belt can operate most efficiently. The sorting belt is 16 feet long and moves horizontally under the main belt from the jaw crusher. The sorted ore discharges into a 100-ton flat-bottom bin. The grizzly undersize enters a spider-type conical trommel 7 feet long and 3 feet in diameter at the discharge end and made of 1/4-inch steel plate punched with 1/2-inch holes. The undersize from the trommel falls directly onto the main belt to the fine-ore bin, whereas the oversize drops into the bin with the sorted ore. Approximately 10 percent of the jaw-crusher discharge is screened out by the trommel and leaves only the coarser material to enter the intermediate crushing unit.

Sorting along this belt has made it possible to handle a greater amount of development ore, which otherwise would have been considered waste. The gangue material consists of mineralized "granite" (quartz-diorite) and quartz. Unless the ore is being drawn from sheared zones in the mine, all the granite is discarded. All the gangue material is deposited into a bin under the picking belt, which is emptied by a mine car running beneath the bin.

A magnetic pulley 24 inches in diameter by 26 inches wide is at the head end of the main belt from the jaw crusher. This removes all the small and intermediate pieces of tramp iron. The large pieces of tramp iron are picked out by the men on the sorting belt or by the man operating the jaw crusher. The magnet obtains its direct current from a motor-generator set under the coarse-ore bin. The 15-kw. generator is direct connected to a 25-horsepower, 2,200-volt induction motor.

Fine Crushing

From the 100-ton storage bin the ore is fed through a side chute onto a 30-inch pan feeder traveling 3.5 feet per minute. It is driven by a 3-horsepower induction motor through a belt drive. The speed of this feeder is controlled by a ratchet and arm and adjusted so that there is a semichoke feed on the crusher. The ore from the feeder discharges onto a screen made of 1/2-inch rods and having a 1-inch opening. The screen oversize goes into a 5N Gates gyratory crusher while the undersize bypasses the crusher, going directly on the No. 2 conveyor. The crusher is set at 3/4-inch and is driven at 600 r.p.m. by a 50-horsepower, 2,200-volt induction motor through a 10-inch belt. The crusher's capacity under its present setting is approximately 20 to 30 tons per hour. The liners and mantle for this crusher are of manganese steel and especially designed for finer grinding than the regular liners. The life of these liners is approximately 1 month, after which time the crusher is dismantled, and the worn places are built up with manganese-steel welding rods. This building-up process is accomplished by a 40-volt, 400-ampere, arc welding machine.

The discharge from this crusher and from the screen drops on an 18-inch belt. This conveyor is 294 feet long, center to center of pulleys, and inclined 19 degrees from the horizontal. This belt, which discharges the ore directly into the fine-ore bin, is at nearly right angles to the long axis of the mill. It is driven at 89 feet per minute through a belt and gear drive connected to a 10-horsepower motor.

Grinding and Classifying

The fine-ore bin is flat-bottomed and has a live-load capacity of 400 tons. The chute from this bin is in the center of the bottom, and the ore passes through a steel hopper onto a 24-inch pan feeder. The feeder is driven by a belt-and-gear reduction at 4-1/2 feet per minute. The amount of feed is controlled by an adjustable gate on the hopper, while the whole feed is shut off by a horizontal rack-and-pinion gate fixed to the bottom of the hopper. The discharge from the feeder drops onto an 18-inch conveyor belt, driven by the motor that drives the pan feeder. This belt is inclined 20 degrees and is 11-1/2 feet long, center to center of drive pulleys. The ore from this belt discharges directly into the drum of a combination drum and scoop feeder attached to the trunnion of the ball mill.

Primary grinding is accomplished in an 8-foot-diameter by a 6-foot-long ball mill. The mill is loaded with 28,000 pounds of 3-inch forged-steel balls and maintained at that weight by adding 600 pounds daily. The shell liners are wave type and cast from chrome-. nickel- molybdenum alloy steel. They are bolted into place in the mill and have an expected life of 40,000 tons original feed. The feed into the ball mill consists of the original feed and the classifier sands. The mill operates at 17 r.p.m. through a gear and pinion driven by a 24-inch, 7-ply transmission belt and connected to a 200-horsepower, 2,200-volt, slip-ring motor.

A 7- by 30-foot Dorr washer converted to a duplex classifier is operated in closed circuit with the ball mill and driven by a 10-horsepower, 220-volt induction motor through an 8-inch belt drive which operates the rakes at 17 strokes per minute on a 2-3/4-inch slope. The sands discharge into a box containing a 12-inch helical-screw conveyor driven by a belt and chain drive connected to the classifier camshaft. The classifier feed and the overflow are maintained as close as possible to 65 and 35 percent solids, respectively, which from experience offers the highest percentage of efficient fine grinding, which is most important in obtaining the best possible recovery. The classifier overflow discharges onto a punched screen, which collects small wood chips.

A regrinding unit has been installed recently to handle the flotation middlings. It consists of a classifier, a ball mill, and a flotation cell, in which approximately 10 tons of middlings per day can be treated resulting in an overall increase in extraction of 5 percent. The unit consists of a 54-inch by 13-foot 6-inch Dorr duplex classifier, which receives the pulp from a 2-inch sand pump handling all the flotation middlings entering the centrally located sump beneath the flotation floor. The classifier operates at 7 strokes per minute on a 1-3/4-inch slope and is belt-driven by a 5-horsepower, 220-volt induction motor. The overflow from this classifier is added to the discharge from the large ball mill, whereas the sands or thickened pulp are fed to a 2-1/2-foot-diameter by 5-foot long ball mill and discharged directly into a 28-inch Fagergren flotation cell. This mill is lined with smooth-type liners and loaded with 1-1/2-inch forged-steel balls. It is driven at 33 r.p.m. by a 10-horsepower, 220-volt motor through a gear and pinion connected to a 6-inch belt. The concentrates from the 28-inch Fagergren cell enter the first cell of the main flotation unit, where they are again floated, and the tailings enter into the middlings cells of the main unit. (See flow sheet.)

FLOTATION

The pulp from the 7- by 30-foot classifier gravitates into a 44-inch Fagergren flotation machine. This machine is equipped with a hard-rubber impeller direct-connected to a 7-1/2-horsepower motor operating at 680 r.p.m. The froth is skimmed off by revolving paddles on each side of the machine, driven through a reduction gear and belt by a 1/2-horsepower motor situated on a platform above the machine. The tailings from this first machine enter a six-cell Kraut flotation machine, while the concentrates gravitate through launders to a 12-foot-diameter thickener. The impellers of the Kraut machine are driven by 7-1/2-horsepower motors through V-belt drives at 775 r.p.m., one motor serving two cells. The froth from this machine is circulated back into the regrinding unit by a 2-inch Krogh sand pump driven at 1,670 r.p.m. through a belt drive connected to a 10-horsepower induction motor. The tailings from the Kraut cells are divided so that approximately 20 tons per 24 hours are run over a No. 6 Wilfley concentrating pilot table operating at 295 strokes per minute and driven through a belt drive connected to a 3-horsepower motor. The following flotation reagents are used:

	Pound per ton
Cresylic acid..........................	.38
Fuel oil...............................	.03
Amyl xanthate..........................	.05
Sodium aerofloat.......................	.03

The xanthates, sodium aerofloat, and cresylic acid are added to the launder that enters the first flotation machine. Cresylic acid is added at intervals to the various cells in the Kraut machine to maintain the froth, while the fuel oil is fed to the middling pump and returned to the regrinding unit. Of all the reagents used, cresylic acid is the most essential. It acts both as a frother and a collector; without its presence the grade of the concentrates is lowered noticeably. The xanthates and sodium aerofloat are important for the collecting of the sulphide minerals present. The amounts added vary, depending upon the mineralization of the ore. Fuel oil acts as a conditioner for the pulp and tends to keep the froth more regular.

Thickening, Filtering, and Drying

Concentrates are dewatered in a wood tank 12 feet in diameter by 11 feet deep, equipped with a Dorr thickener mechanism driven at 1/4 r.p.m. A 3-inch diaphragm pump, operating at 14-strokes per minute, lifts the thickened pulp from the discharge opening of the thickener to an elevation above the tank. The thickener mechanism and the pump are run intermittently. When filtering, the flow from the elevated pump goes into the filter. After filtering, which requires 8 to 12 hours, the rakes in the thickener are raised and the pump and thickener mechanism are shut down. The diaphragm pump and the thickener are driven by the same 5-horsepower motor through belt drives.

The overflow from the peripheral launder in the thickener gravitates through a 2-inch pipe into a 10-foot-diameter wood tank, where further settling takes place. This same tank also acts as a floor drain from the filter and the thickener. The amount of concentrates received in this tank is relatively small, and it is therefore cleaned out only once each month. A 2-inch centrifugal pump direct-connected to a 5-horsepower motor pumps the concentrates from this tank, after the water has been siphoned off, back into the thickener tank against a 20-foot head.

The thickened pulp from the diaphragm pump discharges into a 6- by 6-foot Oliver continuous filter. The drum operates at 1/4 r.p.m., and the pulp is agitated both by paddles and by compressed air through 1/2-inch pipes. During the time of filtering, 20 inches of vacuum are maintained by a 9-1/2- by 8-inch vacuum pump driven by a 10-horsepower motor at 270 r.p.m. Unless the filter cloth becomes torn it will last for 6 months or more.

The cake from the filter, discharged with 15 to 20 percent moisture, drops into a J.H. Day steam-jacketed drier. The screw in this drier is driven at 5-1/2 r.p.m. from the same line shaft as is the filter. A 10-horsepower, 220-volt induction motor is belt-connected to this line shaft, which is directly above the filter. The filter cake is dried in batches. Approximately one ton of filter cake is discharged into the drier per batch, which is its best operating capacity. The boiler is operated at 40 pounds per square inch and circulates the dry steam through the drier. The moisture is reduced to approximately 5 percent in an hour. To lower the moisture much below this percentage would cause a dust loss in handling. The boiler for generating the steam has a rated capacity of 50-horsepower and is fired by wood. A steam heating plant is being contemplated for the mill in which this same boiler will be used.

The finished concentrates, after being discharged from the drier, are then ready for sampling, canning, and weighing. The concentrates are packed in steel cans 16 inches in diameter by 27-1/2 inches high, which, when filled, weigh around 400 pounds each.

Mill Control

All the water used in the mill and mine is pumped from Brown's Branch, a small clear stream 1 mile from the mill. It is pumped directly through a 4-inch cast-iron pipe to a 16,000-gallon wood tank 120 feet above the floor of the ball mill. Except during rainy seasons, the water is very clear and has no bad effect on the flotation circuit.

Densities of the classifier overflow and the ball-mill discharge are checked regularly by the operators. All samples except the tailings are taken by hand at half-hour intervals. The original feed into the ball mill is weighed by taking definite time samples every half hour. A moisture sample of the feed is taken every 8-hour shift. Approximately 100-pound samples of the mill heads are taken during each shift and sent to the assay office, where they are split, crushed, and assayed. The concentrates, middlings, and classifier-overflow samples are taken by hand by cutting the pulp stream with specially made sample cutters.

Each of these samples weighs approximately 10 pounds, and they also are taken every shift. The tailings sample is taken by an automatic sampler designed and built on the property. It cuts the pulp stream every 15 minutes and gives a sample of about 20-pounds per shift. The final concentrates are sampled daily by taking every tenth shovel from the drier discharge and placing it in a sample pile. After all the concentrates from the previous day have been dried and packed, the large sample pile is quartered down and used in making up the shipment sample. A record of the weight and the moisture of these concentrates is kept daily together with the rest of the mill records.

Costs

Summary of Costs

Hog Mountain Gold Mine.

7-month period
(October 1935 - April 1936)

Ore, milled during period - 14,453 tons.
Milling method - flotation.

	Milling costs per ton ore						
					Maintenance		
	Operating labor	Power	Balls	Materials	Labor	Materials	Total
Coarse-ore crushing and conveying....	$0.029	$0.066	-	$0.004	$0.004	-	$0.103
Picking............	.050	-	-	-	.002	-	.052
Fine-ore crushing and conveying....	.021	.059	-	.003	.019	$0.050	.152
Grinding and classifying...........	.054	.260	$0.212	.103	.012	.063	.704
Flotation..........	.062	.048	-	.066	.004	.023	.203
Thickening, filtering and drying...	.020	.008	-	.024	.004	.003	.059
Tailings...........	-	-	-	-	.001	-	.001
Assaying and sampling............	.033	.019	-	.034	.004	.004	.094
Miscellaneous expense............	.092	-	-	.020	.003	-	.115
Total..........	.361	.460	.212	.254	.053	.143	1.483

Summary of costs in units of labor, power, and supplies

Hog Mountain Gold Mine.

7-month period
(October 1935 - April 1936)

Ore milled during period - 14,453

Labor (man-hours per ton of ore milled):	Operating labor	Repair labor	Total
Coarse-ore crushing and conveying	0.23	0.12	0.35
Picking	.08	-	.08
Fine-ore crushing and conveying	.16	.31	.47
Grinding and classifying	.23	.32	.53
Flotation	.23	-	.23
Thickening, filtering, and drying	.17	.06	.23
Tailings	-	.01	.01
Assaying and sampling	.27	-	.27
Other expense	.08	.10	.18
Total	1.45	.92	2.37
Total labor in mill			2.36
Average tons per man-shift in mill			3.39
Labor, percentage of total cost			39.20
Power and supplies:			
Balls (3-inch forged steel), pounds per ton			5.00
Reagents, pounds per ton:			
Cresylic acid			.38
Fuel oil			.03
Amyl xanthate			.05
Sodium aerofloat			.03
Power, kilowatt hours per ton:			
Coarse-ore crushing			4.1
Fine-ore crushing			4.2
Grinding and classifying			17.7
Flotation			3.2
Thickening, filtering, etc			2.1
Assaying and sampling			1.1
Other expense			2.1
Total power			34.5
Water, gallons per ton			700.0

Note. - The water used represents the total consumption for both the mine and the mill. However, the largest proportion of the total is used in the mill.

Tables

TABLE 1. - Typical analysis of mill feed

Au	oz./ton	0.220
Fe	percent	8.28
S	do.	4.55
As	do.	.10
Bi		nil
Ni		nil
Insol	do.	85.60

TABLE 2. - Typical analysis of concentrates

Au	oz./ton	2.75
Ag	do.	.42
Cu	percent	.62
SiO_2	do.	20.24
Fe	do.	42.31
S	do.	28.44
Bi	do.	.08
Pb	do.	.02
Cl	do.	.02
Ni		nil
Zn		nil
MnO	do.	.05

TABLE 3. - Typical analysis of tailings

Au	oz./ton	0.039
Fe	percent	4.2
S	do.	2.5
SiO_2	do.	88.0
Al	do.	3.0

TABLE 4. - Typical analysis of middlings

Au	oz./ton	0.377
Ag	do.	.150
Cu	percent	.37
Fe	do.	43.68
S	do.	13.63
SiO_2	do.	40.5

TABLE 5. - Typical screen analysis

Screen	Ball-mill discharge	Classifier overflow	Classifier sands	Middlings	Tailings	Concentrates
Plus 14..	1.0	-	2.0	-	-	-
" 20..	-	-	3.0	-	-	-
" 30..	-	-	5.0	-	-	-
" 40..	1.0	-	7.0	-	-	-
" 50..	2.5	-	6.0	-	-	-
" 65..	6.5	-	9.0	.25	-	-
" 80..	2.5	5.7	16.0	.25	2.75	-
" 90..	1.5	3.6	5.0	.50	4.00	-
" 100..	5.0	4.6	5.0	1.00	4.00	-
" 150..	35.0	15.6	15.0	6.00	16.00	3.5
" 200..	37.5	18.7	12.0	15.00	20.25	4.5
Minus 200	12.5	51.8	15.0	77.00	53.00	92.0

TABLE 6. - Screen analysis and assays of flotation tailings

	Screen mesh	Percent weight	Assay	Cumulative percent	Value at $35.00/oz.
Plus....	150	30.0	0.045	30.0	$0.472
Plus....	200	17.7	.030	47.7	.175
Minus...	200	52.3	.025	100.0	.468
		100.0	.0314	100.0	$1.115

TABLE 7. - Mill operating data
Feb. 11, 1934 - May 1, 1936

	Over-all period	March 1936 (recent month in period)
Tons crushed............................	50,019	2,749
Tons picked.............................	3,478	302
Tons milled.............................	46,541	2,447
Tons concentrates.......................	2,529	144
Ratio of concentration..................	18.4: 1	17 : 1
Feed assay..............................	.184	.205
Tailing assay...........................	.034	.025
Concentrates assay......................	2.56	2.93
Indicated recovery...............percent	82.3	87.5
Actual recovery................... do.	75.7	84.0
Operating time.................... do.	54.3	70.2
Average operating rate...........tons/day	111	113

Note. - The difference between indicated and actual recovery is due to a mill tie-up, losses during mill shutdowns, and tie-up in the middlings impounded as a separate product.

TABLE 8. - Labor rates

Mine (9 hours per shift):	
Miner	$2.75 - $3.25
Miner's helper	2.00
Mucker	1.75
Timberman	2.50
Timberman helper	2.00
Steel nipper	2.00
Shift boss	3.50 - 4.50
Mill (8 hours per shift):	
Flotation man	2.50
Ball-mill man	2.25
Filter man	2.00
Crusher man	2.00
Pickers	1.00
Roustabout	1.75
Surface (9 hours per shift):	
Steel sharpener	2.75
Steel sharpener's helper	2.00
Topman	1.75
Hoistman	2.75
Machinist	2.75
Pipeman	2.50
Electrician	2.75
Carpenter	2.50
Roustabouts	1.75

The cost of contract work in drifts and crosscuts ranges from $4.50 to $6.00 per foot, depending upon the nature of the ground and the tramming distance. At this price the contractor must furnish both the labor for drilling and mucking and the explosives. Contract work in the raises is a trifle higher.

Contract work has been experimented with in the stopes, the miners being paid 3 to 5 cents per foot of hole drilled, but in general this system has not been satisfactory.

TABLE 9. - Analysis of man-hours

(November 1935 - April 1936)

	Tons	Man-hours per ton
Raising........................tons broken	3,615	4.47
Drifting...................... do.	6,361	3.48
Stoping....................... do.	16,726	2.24
Tramming and mucking..........tons handled	21,466	1.26
Total underground........total tons broken	26,702	2.86
Total milling.................tons milled	13,431	1.40
Picking.......................tons picked	1,987	1.86
Concentrates............tons concentrates	746	1.15
Over-all total labor..........tons handled	21,466	6.33

Note. - The above items take into consideration the total labor used and directly charged to the separate accounts.